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THE APPLICATION OF ROCK BOLT DESIGN METHODS TO A PROPOSED 7 WEST CROSSCUT, PRINCE MINE, SYDNEY COALFIELD, N.S.

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Cape Breton Coal Research Laboratory

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by

T.R.C. Aston* and W.D. Gallant**

ABSTRACT

Rock bolting techniques exhibit considerable potential as primary and/or secondary support systems for use in both temporary and permanent roadways in the Sydney Coalfield, N.S.

The report examines data representative of a proposed 7 West crosscut at Prince Mine, that was input to eight different rock bolt design methods. A comparison of the results indicates that none of the existing methods can be used to design a definitive rock bolt system for use in coal measures strata. Finally, three potential support systems are presented for use in the proposed 7 West crosscut.

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APPLICATION D'UNE MÉTHODE DE CONCEPTION DU BOULONNAGE DANS UNE COUPE TRANSVERSALE PROPOSÉE À LA GALERIE 7 OUEST DE LA MINE PRINCE DU CHAMPS HOUILLER DE SYDNEY EN N.-É.

par

T.R.C. Aston* et W.D. Gallant**

RÉSUMÉ

Les techniques de boulonnage démontrent un important potentiel comme système de soutènement primaire ou secondaire, ou les deux à fois, tant dans les voies de roulage temporaires que permanentes à la houillère de Sydney en N.-É.

Le présent rapport étudie les données représentatives d'une coupe transversale proposée dans la galerie 7 Ouest de la mine Prince. Ces données ont été utilisées avec huit méthodes de conception différentes du boulonnage. Une comparaison des résultats obtenus indiquent qu'aucune des méthodes courantes ne peuvent servir à la conception d'un système de boulonnage définitif réalisable dans les strates d'un gisement houiller. Enfin, trois systèmes de soutènement possibles sont présentés dans le rapport pour leur emploi dans la coupe transversale proposée de la galerie 7 Ouest.

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INTRODUCTION

During 1986 and 1987, discussions between the Cape Breton Development Corporation (CBDC) and the Cape Breton Coal Research Laboratory (CBCRL) revealed considerable interest in the use of rock bolting techniques as a supplementary/primary support system in coal mine roadways and intersections. At a meeting on February 10, 1988 (1,2) it was agreed that a series of field trials should be developed and undertaken and based on the outcome of the results, a more extensive program formulated if appropriate.

One of the first sites identified for a rock bolting trial was a crosscut on the 7 West panel at Prince Mine, Figure 1. However, before a rock bolting trial can be undertaken and because of the lack of previous data on bolting from within the Sydney Coalfield, it was decided that CBCRL should undertake a review of existing rock bolting design criteria and assess their application to the Prince Mine crosscut site.

Initially, the report identifies the various design methods available for determining underground tunnel/roadway support systems. Data representative of the Prince Mine site is then applied to each design method and a comparison of the results made. Finally, three potential rock bolting patterns are presented for consideration by Prince Mine/CBDC engineers.

GEOLOGY - PRINCE MINE, 7 WEST CROSSCUT

Geologically, the coal seam in the vicinity of the 7 West panel is overlain by a very weak, highly slickensided mudstone which in turn is overlain by a massive sandstone unit. The mudstone unit varies in thickness from 0 to 3.6 m, while the top of the sandstone unit is approximately 11.5 m above the seam. It is very difficult to predict accurately the thickness of the mudstone unit without undertaking an extensive drilling program.

Several roof falls have occurred in roadways overlain by the mudstone in the vicinity of the 5 West panel and these extended up to a height of about 3.6 m

A geological cross section along the 6 West bottom level is shown in Figure 2.

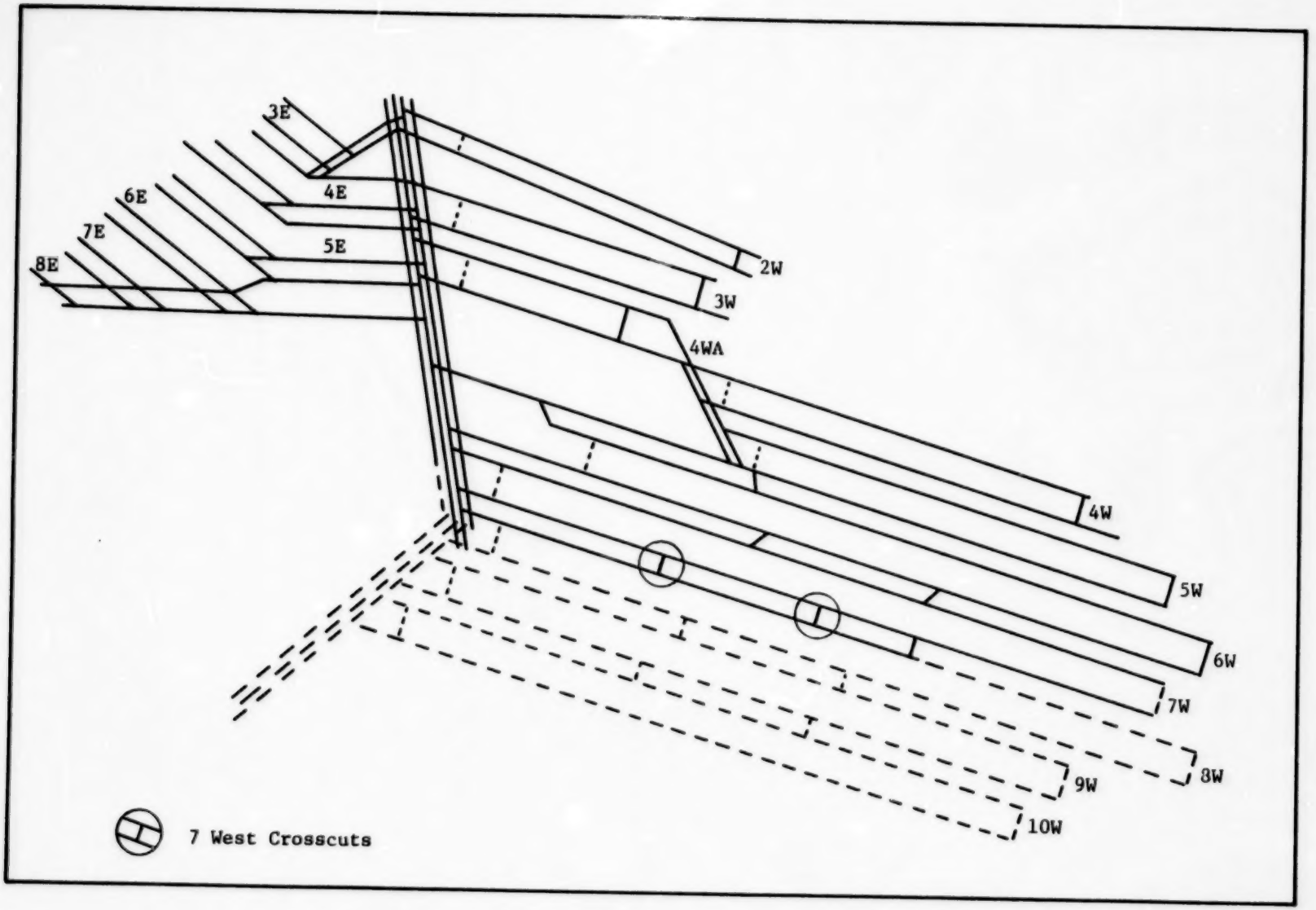

Figure 1 - Schematic plan showing the Prince Mine workings

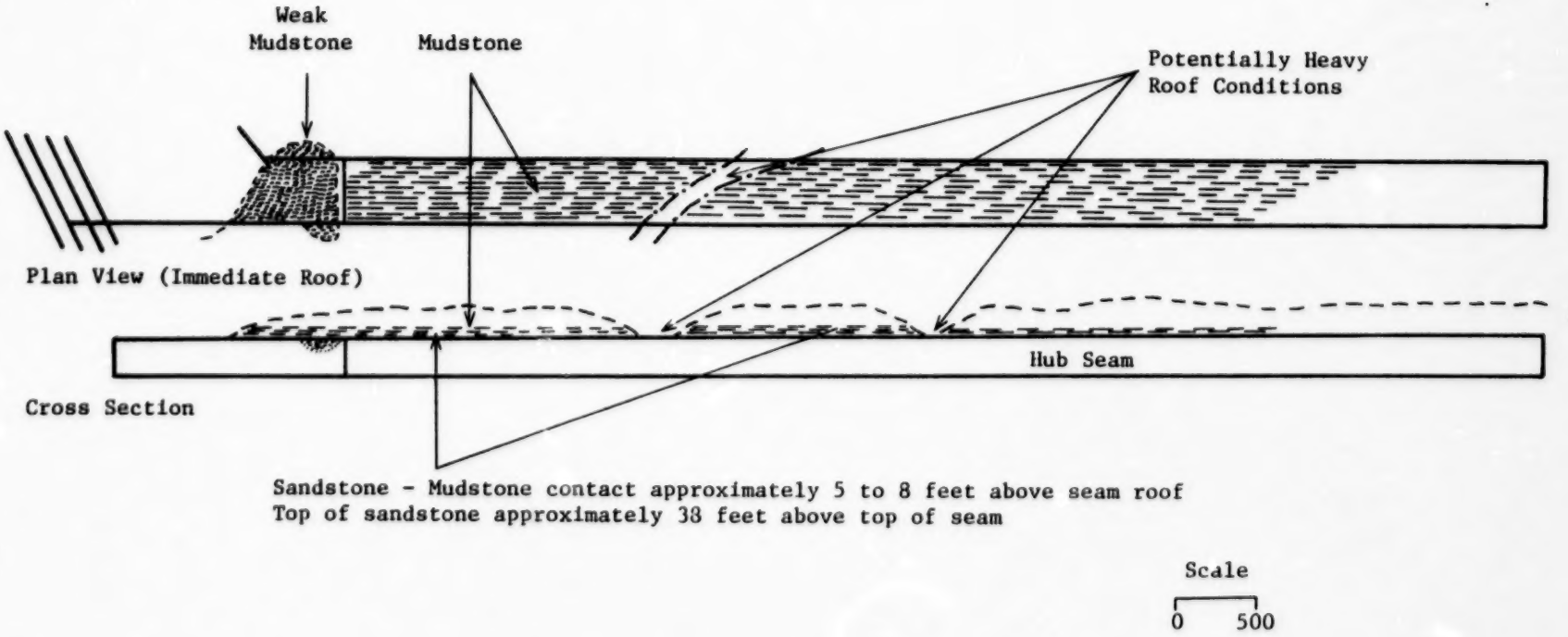

Figure 2 - Geological conditions along 6 West bottom level, Prince Mine

PROPOSED CROSSCUT DIMENSIONS

The proposed 7 West crosscut would be driven inseam, with an approximate length of 75 m. At Prince Mine, conventional crosscuts are supported by rectangular steel sets on 0.9 m spacings. The roof span is projected to be around 4.27 m (14') with a leg height of 1.98 to 2.13 m (6½' to 7'). The seam height is thought to be 2.5 m.

ROCK MASS CLASSIFICATION SYSTEMS

Two rock mass classification systems are commonly used to provide an assessment of the factors likely to influence the stability of an underground excavation: Geomechanics and Q-System. It should be noted that both systems were derived from data predominantly obtained from permanent civil engineering tunnels and their application to temporary coal mining roadways is therefore uncertain.

A description of both the Geomechanics and Q-System classification systems is beyond the scope of this report and can be found elsewhere (3,4).

GEOMECHANICS SYSTEM

The Geomechanics System uses five basic rock mass parameters. A points rating is assigned to a series of parameters and the sum of the parameter ratings is called the RMR (Rock Mass Rating) value. The RMR value is then used to assess the stand-up time of the underground excavation (3,4).

Based on information provided by CBDC geologists, the following ratings were derived for the classification parameters:

Parameter (for Mudstone)	Value	Rating
Rock Strength (MPa)	5-25	2
RQD	<25%	3
Fracture Spacing	<60 mm	5
Discontinuity Condition		10
Groundwater Conditions	damp	10
	Basic Rock Mass Value	30
Joint Orientation	Very Unfavourable	-12
		18

Using Figure 3 and assuming an unsupported roof span of 4.27 m, the estimated stand-up time of the 7 West crosscut would be <1 hour (5).

An RMR value of 18 indicates a Class IV or V rock mass (poor to very poor rock type) (3):

	Class IV	Class V
Average Stand-up Time	10 hrs for 2.5 m span	30 min for 1 m span
Rock Mass Cohesion	100-200 kPa	<100 kPa
Rock Mass Friction Angle	15° - 25°	<15°

Table 1 shows adjustments that can be made to the Geomechanics System for mining applications.

If,

A_B = 1.0 (crosscut machine driven)
A_S = 0.9 (in situ stress regime 2:1)
S = 0.9 (mudstone highly slickensided)

$$RMR_{new} = RMR \times A_B \times A_S \times S \qquad \text{Eq. 1}$$
$$= 18 \times 1.0 \times 0.9 \times 0.9$$
$$= \leq 15$$

Adjustment for a mining application gives a new RMR value of ≤15, which still indicates an unsupported roof stand-up time of <1 hour.

Finally, the potential support load can be estimated using the Geomechanics System (4,6) and Figure 4 gives the estimated rock load per unit length of the 7 West crosscut as 600-700 kN/m.

Q-SYSTEM

The Q-System is based on a numerical assessment of rock mass quality using six parameters: 1) RQD (Rock Mass Quality), 2) number of joint sets, 3) roughness of the most unfavourable joint or discontinuity, 4) degree of alteration or filling along the weakest joint, 5) water inflow and 6) stress condition. The system was developed from the analysis of some 200 tunnel case histories from Scandinavia by Barton, Lien and Lunde at the Norwegian Geotechnical Institute (7,8).

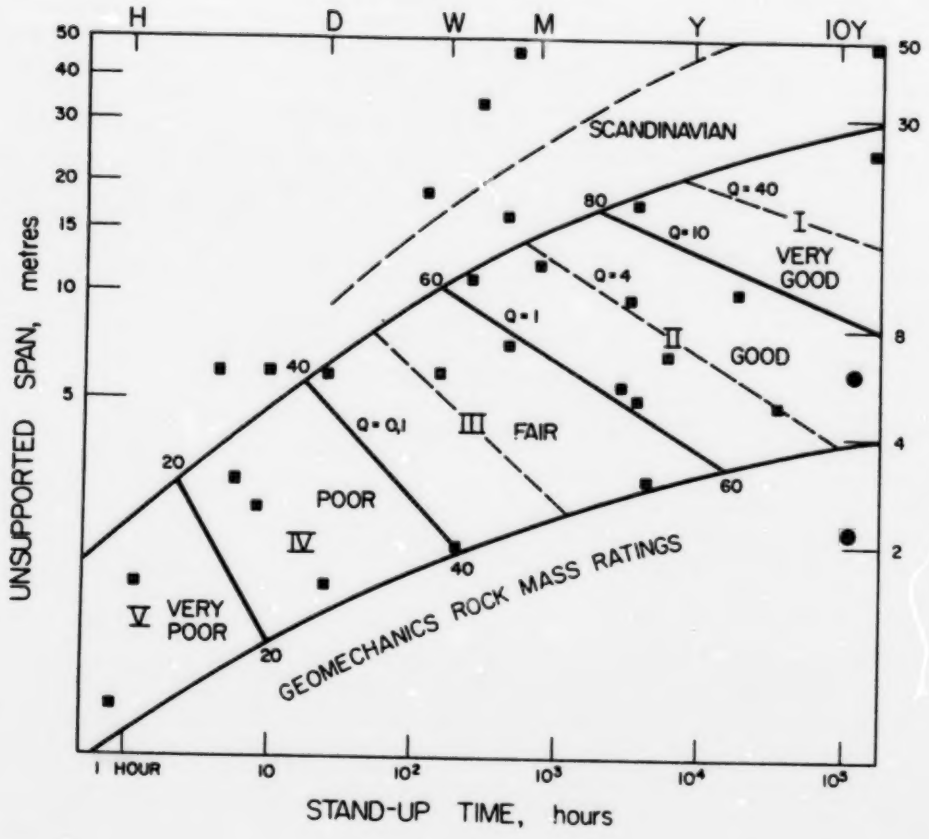

Figure 3 - Relationship between the RMR-value and stand-up time of an unsupported underground excavation span (after (5))

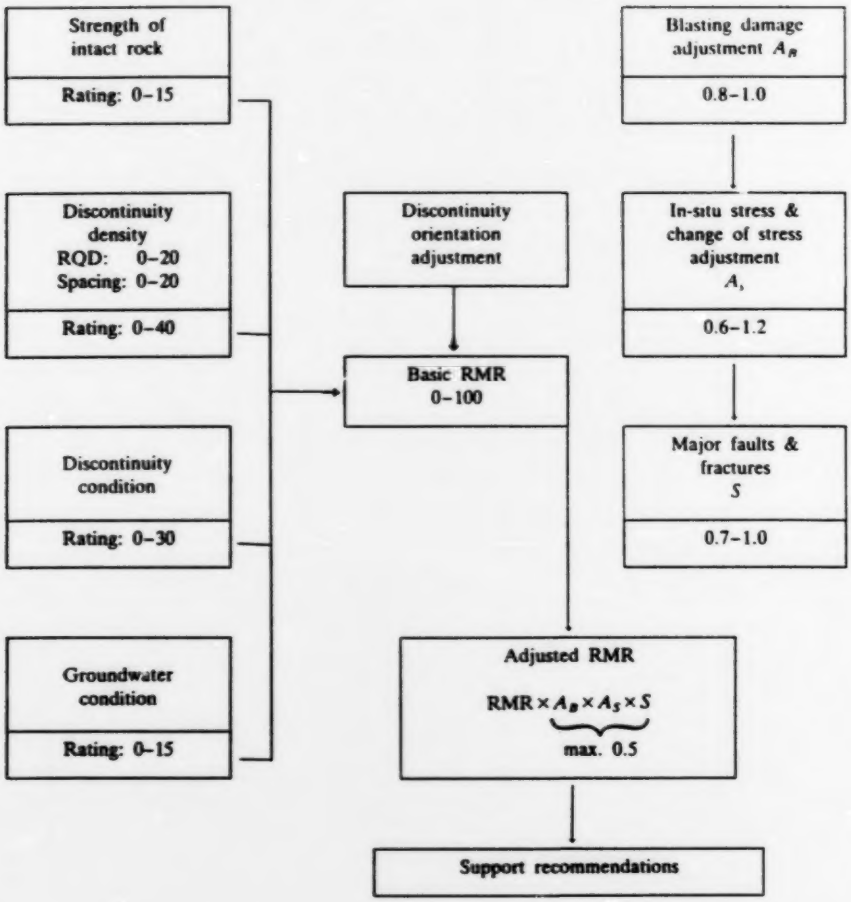

Table 1 - Mining application adjustments for the Geomechanics Classification System (after (4))

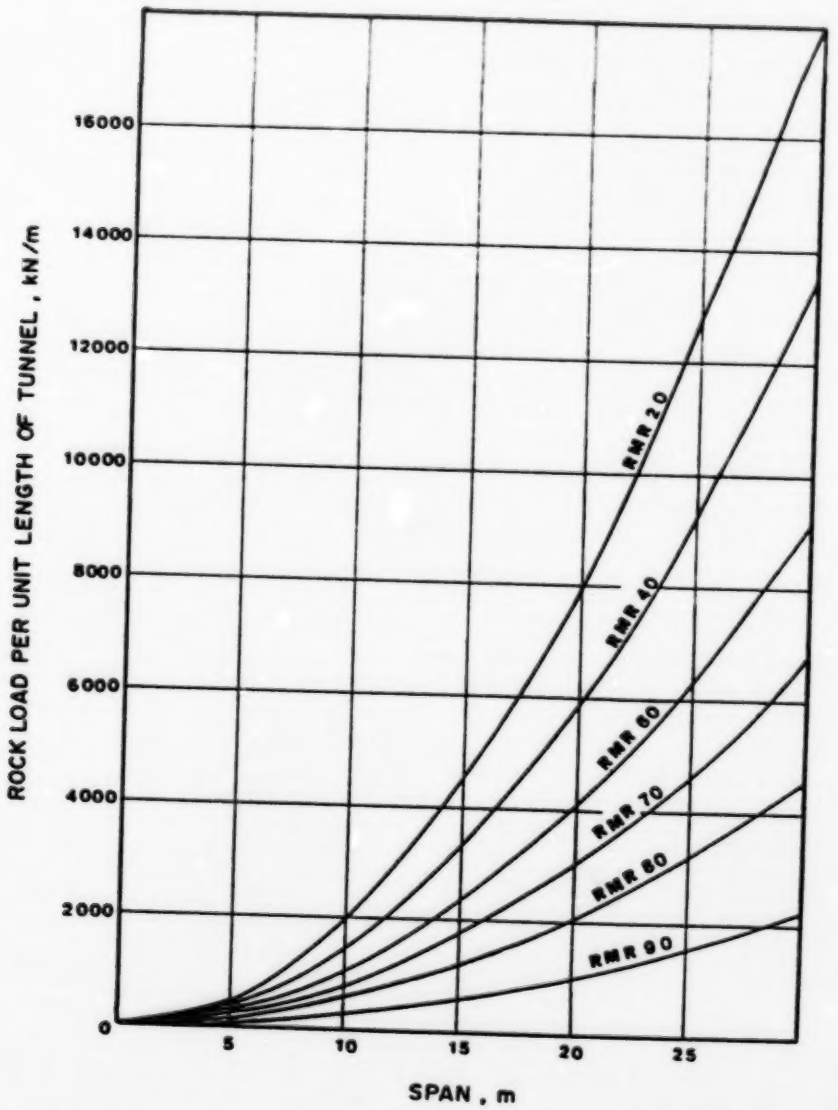

Figure 4 - Variation of rock load as a function of roof span for different rock classes in the Geomechanics Classification System (after (6))

The six parameters are grouped into three quotients to give an overall rock mass quality, Q, (4,7,8):

$$Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \times \frac{Jw}{SRF} \qquad \text{Eq. 2}$$

where, Q = Overall Rock Mass Quality
RQD = Rock Quality Designation
Jn = Joint Set Number
Jr = Joint Roughness Number
Ja = Joint Alteration Number
Jw = Joint Water Reduction Number
SRF = Stress Reduction Factor

Based on information provided by CBDC geologists, the following ratings were derived for the classification parameters:

RQD = 10	Ja = 3.0
Jn = 2.0	Jw = 1
Jr = 1.0	SRF = 10

$$Q = \frac{10 \times 1 \times 1}{20 \times 3 \times 10} = \frac{10}{600} = 0.017$$

To relate the 'Q' value to the behaviour and support requirements of an underground excavation, it is necessary to determine the Equivalent Dimension (D_e), which is calculated using Equation 3 (3,7,8):

$$D_e = \frac{\text{Excavation span, diameter or height (m)}}{\text{Excavation Support Ratio (ESR)}} \qquad \text{Eq. 3}$$

The numerical value of the excavation support ratio (ESR) is related to the intended use of the excavation and the safety factor required. ESR values for a variety of excavation catagories are given in Table 2 (3,9).

	Excavation Category	ESR
A.	Temporary mine openings	3-5
B.	Permanent mine openings, water tunnels for hydropower (excluding high pressure penstocks) pilot tunnels, drifts and headings for large excavations	1.6
C.	Storage rooms, water treatment plants, minor road and railway tunnels, surge chambers, access tunnels	1.3
D.	Power stations, major road and railway tunnels, civil defence chambers, portals, intersections	1.0
E.	Underground nuclear power stations, railway stations, sports and public facilities, factories	0.8

Table 2 - Excavation support ratio (ESR) related to the intended use of the excavation (after (9))

The relationship between the 'Q' value and the equivalent dimension D_e of an excavation which will stand unsupported is given in Figure 5 (3,8). Also plotted on Figure 5 are the three equivalent dimension values (Table 2) that can be derived for temporary mine openings.

Case 1 ESR = 3 D_e = 1.42
Case 2 ESR = 4 D_e = 1.07
Case 3 ESR = 5 D_e = 0.85

Cases 1 to 3 show that a support system must be installed in the roadway.

Using Figure 5, where Q = 0.017, the maximum equivalent dimension, D_e, for an unsupported excavation is 0.45 m. The maximum unsupported roof spans that could therefore be considered for the 7 West crosscut are, using Equation 4:

$$\text{Span (m)} = \text{ESR} \times D_e \qquad \text{Eq. 4}$$

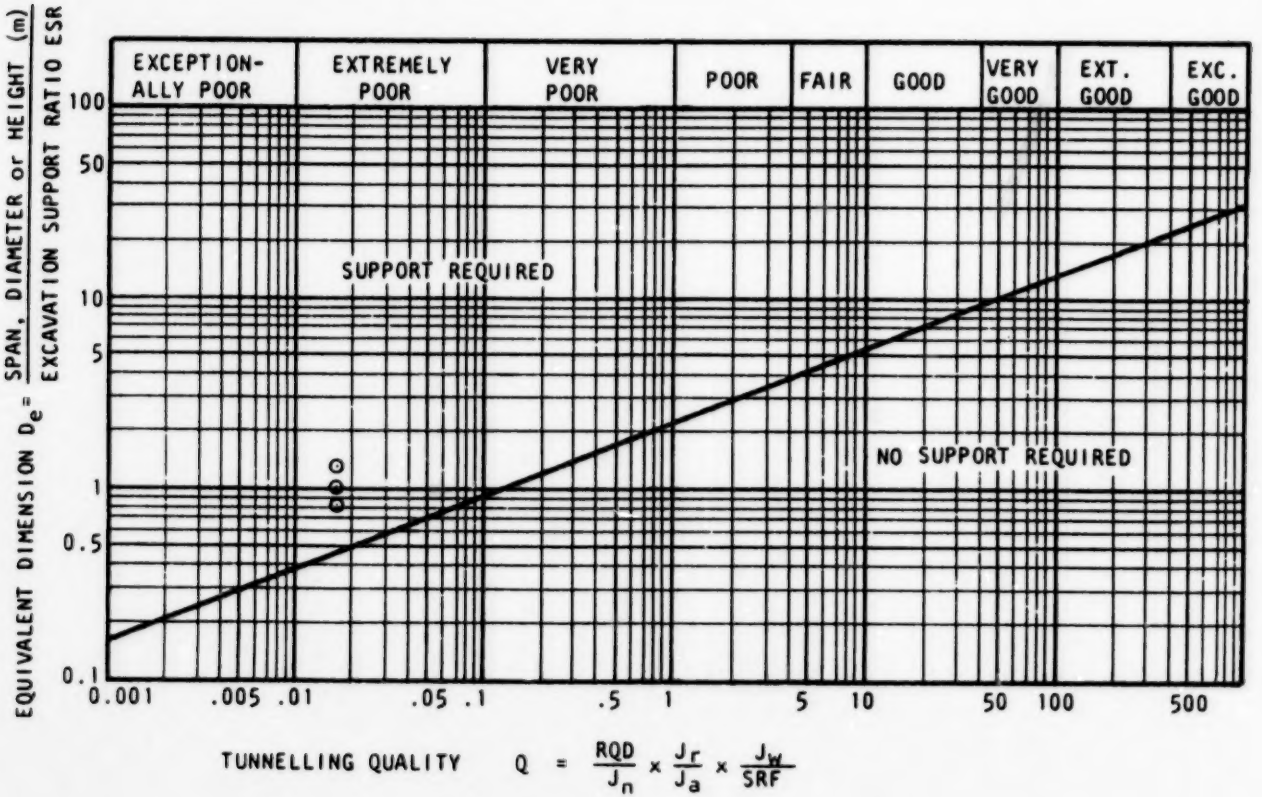

Figure 5 - Relationship between the maximum equivalent dimension (D_e) of an unsupported underground excavation and the rock mass quality (Q) (after (8))

Case 1 = 3 x 0.45 = 1.35 m
Case 2 = 4 x 0.45 = 1.80 m
Case 3 = 5 x 0.45 = 2.25 m

ROCK BOLT SYSTEM DESIGN METHODS

The design of rock bolt systems for the support of excavations can be divided into two main categories: 1) empirical methods and 2) analytical methods. It is therefore proposed to apply the 7 West crosscut conditions to a variety of empirical and analytical design methods.

EMPIRICAL METHODS

1) U.S. Corps of Engineers

A simplified system for rock bolt reinforcement design has been formulated by the U.S. Corps of Engineers (3,10). The empirical rules, given in Table 3, can be used to determine the minimum reinforcement required during the preliminary design stages.

Using Table 3, the minimum length of the bolts required for the 7 West crosscut are:

1) From b): 3 x thickness of potentially unstable rock
3 x 3.66* m (12') = 11 m

*Value determined from height of roof falls experienced on the 5 West district.

2) From c): Spans <6 m: 0.5 x span
0.5 x 4.27 = 2.14 m

Also, the minimum spacing requirement is 0.9 to 1.2 m, while the maximum spacing is:

1) From b): 0.5 x bolt length
0.5 x 2.14 m = 1.07 m

2) From c): 2.0 m

Parameter	*Empirical rule*
Minimum length and maximum spacing	
Minimum length	Greatest of: (a) 2 x bolt spacing (b) 3 x thickness of critical and potentially unstable rock blocks (Note 1) (c) For elements above the springline: spans <6 m: 0.5 x span spans between 18 and 30 m: 0.25 x span spans between 6 and 18 m: interpolate between 3 and 4.5 m (d) For elements below the springline: height <18 m: as (c) above height >18 m: 0.2 x height
Maximum spacing	Least of: (a) 0.5 x bolt length (b) 1.5 x width of critical and potentially unstable rock blocks (Note 1) (c) 2.0 m (Note 2)
Minimum spacing	0.9 to 1.2 m
Minimum average confining pressure	
Minimum average confining pressure at yield point of elements (Note 3)	Greatest of: (a) Above springline: *either* pressure = vertical rock load of 0.2 x opening width or 40 kN/m² (b) Below springline: *either* pressure = vertical rock load of 0.1 x opening height or 40 kN/m² (c) At intersections: 2 x confining pressure determined above (Note 4)

Notes:

1. Where joint spacing is close and span relatively large, the superposition of two reinforcement patterns may be appropriate (e.g. long heavy elements on wide centres to support the span, and shorter, lighter bolts on closer centres to stabilise the surface against ravelling).
2. Greater spacing than 2.0 m makes attachment of surface support elements (e.g. weldmesh or chain link mesh) difficult.
3. Assuming the elements behave in a ductile manner.
4. This reinforcement should be installed from the first opening excavated prior to forming the intersection. Stress concentrations are generally higher at intersections and rock blocks are free to move toward both openings.

Table 3 - Empirical design guidelines (after (3,10))

2) Geomechanics System

Using Table 4, which shows the modified geomechanics classification for the support of mining excavations (3,11) where RMR = 18 and RMR_{new} = ≤15, the recommended support requirements are:

f) - Patterned grouted bolts at 0.75 m collar spacing and shotcrete 100 mm thick.

h) - Massive concrete 450 mm thick with patterned grouted bolts at 1 m spacing if stress changes not excessive.

i) - Grouted bolts at 0.75 m collar spacing if reinforcing potential is present; 100 mm reinforced shotcrete and yielding arches as a repair technique if stress changes not excessive.

j) - Stabilise with rope cover support and massive concrete 450 mm thick if stress changes not excessive.

3) Q-System

Figure 6 shows the unsupported span of some 30 cases of permanently unsupported excavations plotted against the rock mass quality, Q, (9). However, there is no way of determining how close the excavations are to failure and it is therefore likely to always provide a conservative estimate of the unsupported excavation span (9,12). Equation 5 defines the upper limit line shown in Figure 6:

$$\text{Span} = 2 \times Q^{0.66} \qquad \text{Eq. 5}$$

Figure 7 shows the maximum unsupported spans for different excavation support ratios, ESR, plotted against rock mass quality, Q. Equation 5 defines the lines plotted on Figure 7, (9):

$$\text{Span} = 2 \times \text{ESR} \times Q^{0.4} \qquad \text{Eq. 6}$$

Alternatively, the critical value of Q for a given excavation span is expressed in Equation 7:

$$Q = \left(\frac{\text{Span}}{2 \times \text{EMR}}\right)^{2.5} \qquad \text{Eq. 7}$$

Adjusted ratings	Original rock mass ratings, *RMR*									
	90—100	80—90	70—80	60—70	50—60	40—50	30—40	20—30	10—20	0—10
60—100										
50—60		*a*	*a*	*a*	*a*					
40—50			*b*	*b*	*b*	*b*				
30—40				*c,d*	*c,d*	*c,d,e*	*d,e*			
20—30					*g*	*f,g*	*f,g,j*	*f,h,j*		
10—20						*i*	*i*	*h,i,j*	*h,j*	
0—10							*k*	*k*	*l*	*l*

a) Generally no support but locally joint intersections might require bolting.
b) Patterned grouted bolts at 1 m collar spacing.
c) Patterned grouted bolts at 0.75 m collar spacing.
d) Patterned grouted bolts at 1 m collar spacing and shotcrete 100 mm thick.
e) Patterned grouted bolts at 1 m collar spacing and massive concrete 300 mm thick and only used if stress changes not excessive.
f) Patterned grouted bolts at 0.75 m collar spacing and shotcrete 100 mm thick.
g) Patterned grouted bolts at 0.75 m collar spacing with mesh reinforced shotcrete 100 mm thick.
h) Massive concrete 450 mm thick with patterned grouted bolts at 1 m spacing if stress changes are not excessive.
i) Grouted bolts at 0.75 m collar spacing if reinforcing potential is present, and 100 mm reinforced shotcrete, and then yielding steel arches as a repair technique if stress changes are excessive.
j) Stabilise with rope cover support and massive concrete 450 mm thick if stress changes not excessive.
k) Stabilise with rope cover support followed by shotcrete to and including face if necessary, and then closely spaced yielding arches as a repair technique where stress changes are excessive.
l) Avoid development in this ground otherwise use support systems *j* or *k*.

Supplementary notes

I The original Geomechanics Classification as well as the adjusted ratings must be taken into account in assessing the support requirements.
II Bolts serve little purpose in highly jointed ground and should not be used as the sole support where the joint spacing rating is less than 6.
III The recommendations contained in Table 7 are applicable to mining operations with stress levels less than 30 MPa.
IV Large chambers should only be excavated in rock with adjusted total classification ratings of 50 or better.

Table 4 - Modified Geomechanics Classification Guide for the support of mining excavations (after (3,11))

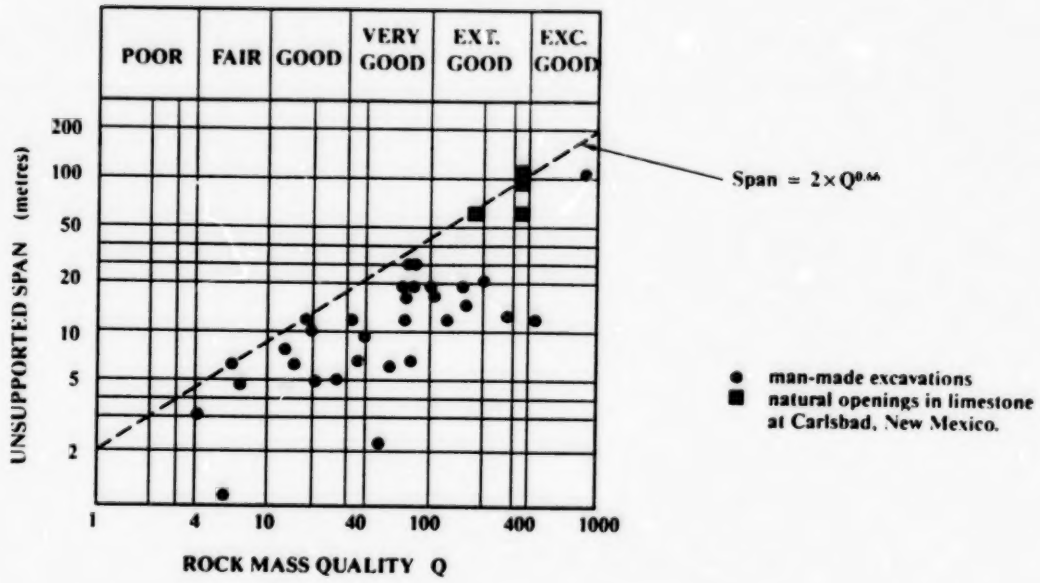

Figure 6 - Man made and natural, unsupported excavations in different quality rock masses (after (9))

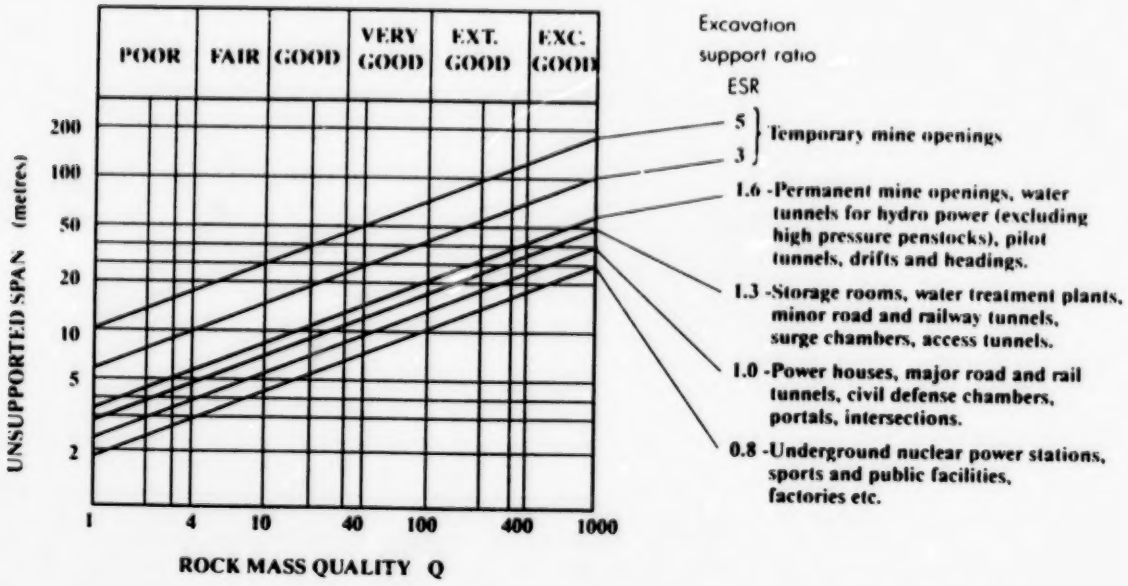

Figure 7 - Recommended maximum unsupported excavation spans for different rock mass quality (Q) and ESR-values (after (9))

Using Equation 5, the calculated unsupported roof span for the 7 West crosscut, where Q = 0.017 is 0.14 m.

Alternatively, using Equation 6 for the three ESR values available for temporary mine openings gives calculated unsupported roof spans of:

Case 1 ESR = 3 Span = 1.2 m
Case 2 ESR = 4 Span = 1.6 m
Case 3 ESR = 5 Span = 2.0 m

Using Equation 7, the critical 'Q' values for the 7 West crosscut (span = 4.27 m) can be calculated for the three ESR values:

Case 1 ESR = 3 Q = 0.427
Case 2 ESR = 4 Q = 0.208
Case 3 ESR = 5 Q = 0.120

Figure 8 shows the rock mass quality, Q, against Excavation Support Ratio, ESR, onto which has been superimposed 38 categories of support measures suggested for use in the anticipated conditions (8). The suggested support measures for the 38 categories are listed in Appendix A.

Using Q = 0.017 and D_e values of 1.42, 1.07 and 0.85 it is apparent from Figure 8 that the 7 West crosscut conditions will be extremely poor, for which the recommended support type is No. 33:

Support Category	RQD Jn	Jr Ja	Span ESR	Support Type	Notes
33*	≥2	-	-	B (tg) 1 m +	IX
				S (mr) 2.5-5 cm	
	<2	-	-	S (mr) 5-10 cm	IX
	-	-	-	S (mr) 7.5-10 cm	VIII,X

*Insufficient case records available for confident prediction of support requirements.

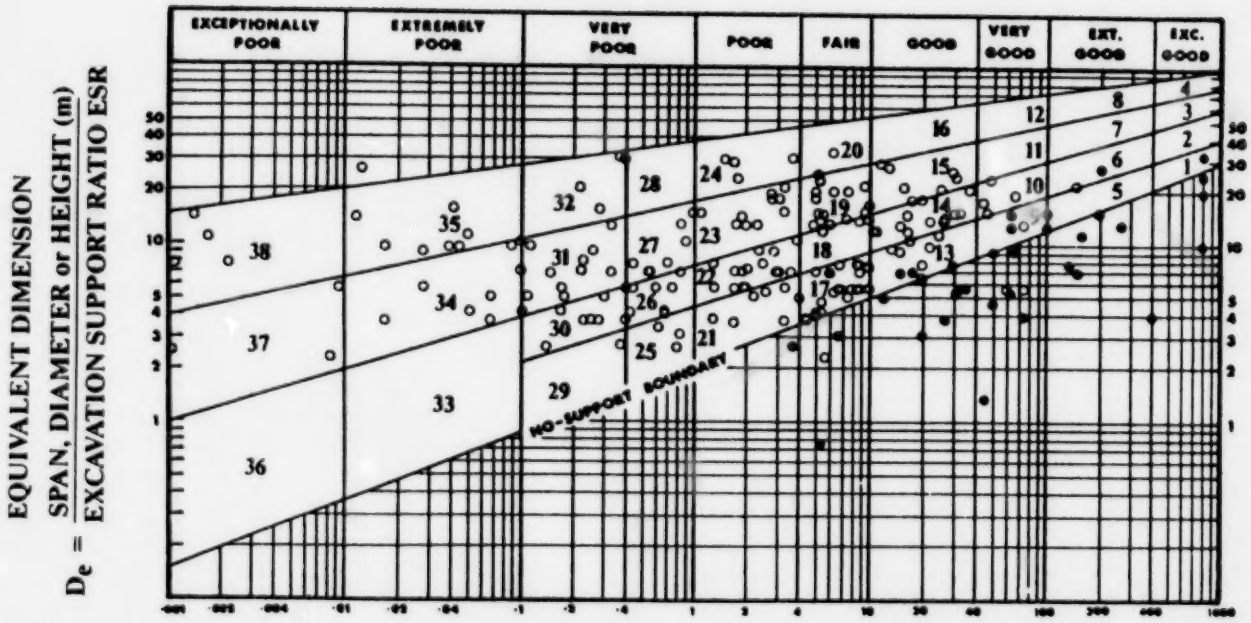

Figure 8 - Recommended support guide for different rock mass quality (Q) and ESR-values. Support for Categories 1 to 38 are listed in Appendix A (after (8))

B - Systematic Bolting

(t_g) - Tensioned (expansion shell type in competent rock masses, grouted post-tensioned in very poor quality rock masses)

S - Shotcrete

(mr) - Mesh reinforced

Notes:

VIII - Cases involving swelling, for instance montmorillonite clay (with excess of water). Room for expansion behind the support is used in cases of heavy swelling. Drainage measures are used where possible.

IX - Cases not involving swelling clay or squeezing rock.

X - Cases involving squeezing rock. Heavy rigid support is generally used as a permanent support.

4) French System

An empirical approach to rock bolt design has been developed for use in French coal mines. Essentially, a matrix approach was developed which relates the roadway properties to the bolting parameters. To derive a required pattern it is necessary to ascertain the existing conditions and work through the matrix (13). The French design matrix is shown in Table 5.

1) Rock Type

This is a simplistic classification designed to characterise the strata immediately surrounding the roadway. Essentially, a judgement is made on whether the rock is solid or liable to slab.

2) Stress Field in the Surrounding Strata

This information should be obtained from detailed field investigations. Two distinct categories have been defined: superficial and deep. The superficial category concerns the strata behaviour to a depth of 1 m around the roadway. The deep category concerns the overall deep seated rock mass behaviour surrounding the roadway.

3) Time Effects

Two simple parameters have been categorised: stable deformation and delayed deformation. These parameters refer to the rheological

Type of rock and strata	Result of stress field in strata	Effect on time	Bolting Parameters: Type of Anchorage	Length	Diameter	Density	Lagging
Homogeneous or slighty fractured	Deep and superficial stability	Stabilised deformation	Support not necessary				
		Delayed deformation	Light point	Short	Small	Low	Light shotcrete
	Deep stability and superficial instability	Stabilised deformation	Any	Short	Small	Medium	Light
		Delayed deformation	Light or strong point	Short	Small	Medium	Light continuous
	Deep and superficial instability	Stabilised deformation	Strong point or full column	Medium to long	Medium	High	Heavy
		Delayed deformation	Strong point	Medium	Medium	High	Heavy
Stratified and hardly vertically	Deep and superficial stability	Stabilised deformation	Support not necessary				
		Delayed deformation	Light point	Short	Small	Low	Light shotcrete
	Deep stability and superficial instability	Stabilised deformation	Any	Medium	Small	Medium	Light
		Delayed deformation	Light or strong point	Medium	Medium	Medium	Light continuous
	Deep and superficial instability	Stabilised deformation	Strong point or full column	Long	Medium	High	Heavy
		Delayed deformation	Strong point	Long	Large	High	Heavy
Irregular lenticular or fractured in several directions	Deep and superficial stability	Stabilised deformation	Point or full column	Short	Small	Medium	Light continuous
		Delayed deformation	Point or full column	Short	Small	Medium	Light continuous
	Deep stability and superficial instability	Stabilised deformation	Full column	Medium	Medium	Medium	Light
		Delayed deformation	Strong point or full column	Medium	Medium	Medium	Heavy
	Deep and superficial instability	Stabilised deformation	Full column	Long	Large	High	Heavy
		Delayed deformation	Strong point or full column	Long	Large	High	Heavy

Table 5 - French matrix for rock bolt support design

properties of the strata and mining factors such as adjacent workings which might cause time dependent deformation. Detailed field investigations should be done to determine these properties.

4) Anchor Type

This column is self-explanatory, although in France there are two types of point anchors: 1) where the bolt fails before the bond (strong) and 2) where the bond normally fails first (weak).

5) Length

Short bolts are about 1 m long. Medium bolts are about one-third the width of the structure, normally 1.8 to 2.4 m. Long bolts are one-third to one-half of the structure width.

6) Diameter

Small diameters are less than 20 mm. Medium is between 20 and 25 mm, while anything greater than 25 mm is considered large.

7) Density

Low density is less than 0.75 bolts/m^2, medium is 0.75 to 1.25 bolts/m^2 and high is over 1.25 bolts/m^2.

8) Lagging

Light discontinuous lagging consists of steel strips or beams. Light continuous lagging consists of welded wire mesh. Heavy lagging consists of heavy duty wire mesh either with or without shotcrete.

9) Other Parameters

Although the matrix includes the major factors likely to affect the support design, additional factors which should also be considered are: bolt tension, steel type, bolt surface and corrosion resistance.

It should be noted that the matrix is only intended for the initial design studies. The final system chosen should be monitored with respect to

both technical and economic feasibility. 'Fine-tuning' should therefore be undertaken during field trials in case local conditions necessitate revisions.

Using Table 5 in conjunction with the known 7 West crosscut conditions, the suggested bolting system is:

1) Irregular Lenticular or Fractured in Several Directions
2) Deep and Superficial Instability
3) Delayed Deformation
4) Anchor Type - Strong Point or Full Column
5) Length - Long (1.4 to 2.1 m)
6) Bolt Diameter - Large (over 25 mm)
7) Bolt Density - High (over 1.25 bolts/m^2)
8) Lagging - Heavy (heavy duty wire mesh - with/without shotcrete)

ANALYTICAL METHODS

In sedimentary formations, such as the Coal Measures, the failure mechanism of an underground opening is normally controlled by bedding planes which exist within the surrounding rock mass. Rock bolting techniques can be used to reinforce the roof beam and form a more competent structural member (3,14).

1) Bedded Strata with an Overlying Competent Unit

Where a weak strata unit is overlain by a more competent member, rock bolts can be used to anchor the weaker unit to the more competent unit. The number of rock bolts required is determined from the dead weight of the weak unit, using Equation 8 (3,14):

$$W = f \times s \times c \times h \times e \qquad \text{Eq. 8}$$

where W = Weight of rock to be supported by a single bolt.
f = Safety Factor, normally $1.5 \leq f < 3$. The value of 'f' depends upon the damage resulting from falling rock and on whether permanent or temporary reinforcement is required.
s = Spacing, perpendicular to excavation axis.

c = Bolt Spacing, along excavation axis.
h = Thickness of unstable rock layer.
e = Rock Density, usually 2.5 t/m^3.

Using information relevant to the 7 West crosscut, the following values can be input to Equation 8:

f = 2.0
s = 1.0 m
c = 1.0 m
h = 3.66 m
e = 2.5

W = 2.0 x 1.0 x 1.0 x 3.66 x 2.5 = 18.3 tons

Although a 4 m bolt appears acceptable, because of the unknown thickness of the mudstone overlying the 7 West crosscut a 4.5 m bolt might be more appropriate.

2) Bedded Strata

In a bedded rock mass with horizontal laminations, tensioned rock bolts can be used to reduce the amount of bed separation and increase the roof stability. Figure 9 shows a nomogram developed to assist with the design of bolting patterns in horizontally bedded strata and its use is described in steps 1 to 5.

1) Estimate the mean thickness, 'e' of the individual layers of rock. Choose a rock bolt length which permits the bolt to be anchored in a layer of sufficient thickness and strength for good anchoring of the bolt.
2) In pull-out tests, the correct level of tensioning is determined, (the tension applied to the bolts should be approximately 0.6 x the tensile strength of the bolt).
3) The number of bolts per set (in each row), perpendicular to the axis of the opening is estimated. If joints are present between the system of bolts, (joints that are likely to cause rock falls),

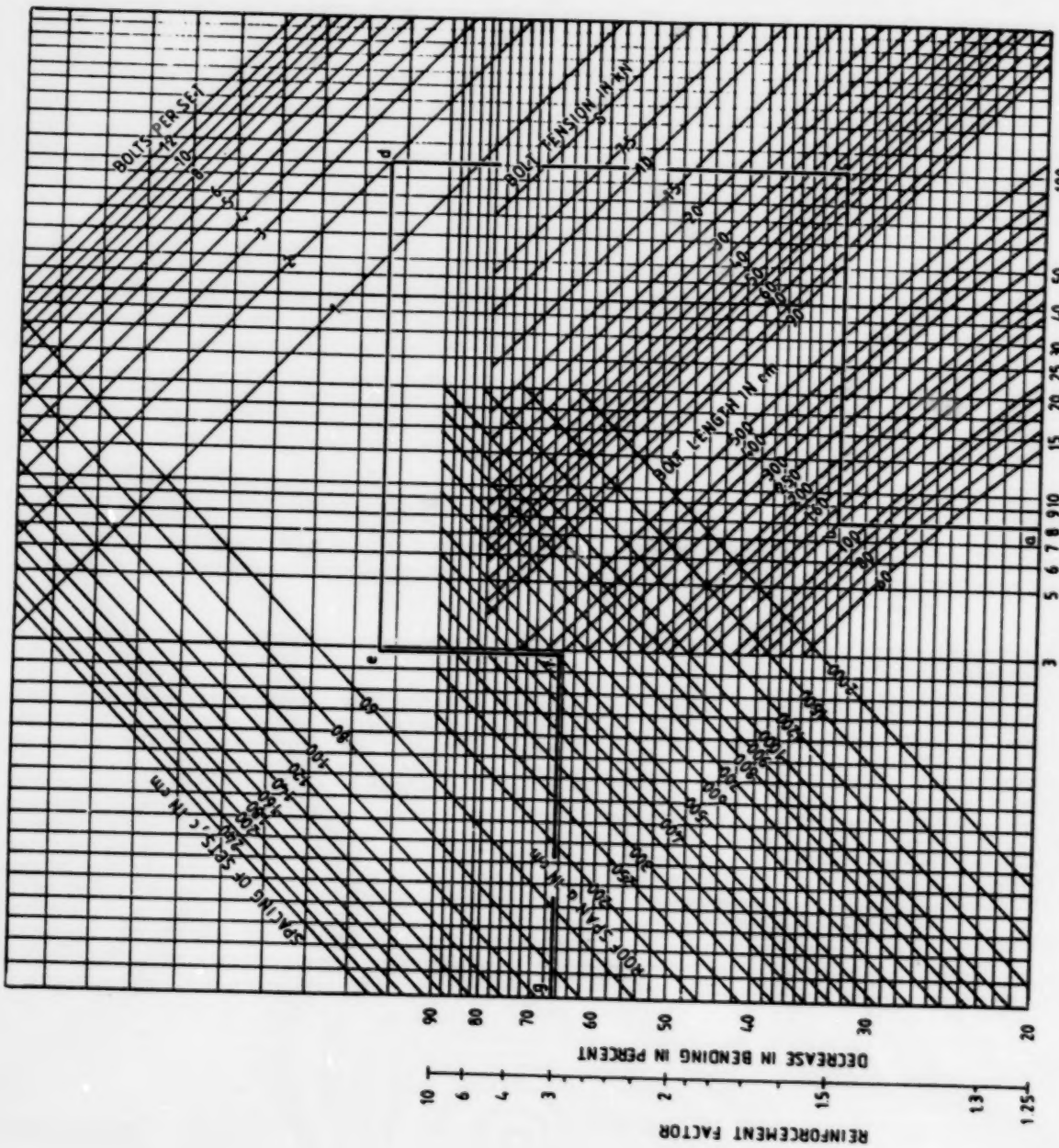
BOLTS PER SET
BOLT TENSION IN kN
BOR LENGTH in cm
BED THICKNESS, e IN cm
DECREASE IN BENDING IN PERCENT
REINFORCEMENT FACTOR
SPACING OF SETS c IN cm

Figure 9 - Design chart for rock bolt reinforcement of horizontally-bedded weak rock (after (14))

the distance, 's' between the bolts should be adjusted accordingly, ($s \leq 3\ e$).

4) The spacing 'c' between sets of bolts along the axis of the opening is selected. Normally the spacing 'c' is of the same order of magnitude as the distance 's' between the bolts.

5) By using the above information together with the known span of the opening in the chart, it is possible to determine the Reinforcement Factor (which should not be less than 2). The Reinforcement Factor is a measure of the percentage decrease in the sag (bending) of the roof that is obtained from rock bolting. A factor of 2 corresponds to a reduction in the sag by half, compared with an unsupported roof.

Three potential cases, relating to the 7 West crosscut are now examined:

Case 1: A 90% reduction in roof sag (reinforcement factor of 10) is required for a 450 cm roof span supported by rock bolt arrays on 1 m spacings with four bolts per array. If the bolts have a tension or equivalent tension of 80 kN, the required bolt length is 1.6 m for a bed thickness of 5 cm.

Case 2: A 67% reduction in roof sag (reinforcement factor of 3) is required for a 450 cm roof span supported by rock bolt arrays on 1 m spacings with four bolts per array. If the bolts have a tension or equivalent tension of 80 kN, the required bolt length is 0.65 m for a bed thickness of 5 cm.

Case 3: If the bed thickness is 5 cm and 2.5 m bolts with a tension or equivalent tension of 80 kN are used in arrays of 1 m spacing with four bolts per array across a roof span of 450 cm. The estimated decrease in roof sag will exceed 90% (reinforcement factor in excess of 10).

3) Rock Arch Concept

If an excavation with a curved roof is driven through a jointed rock mass, a 'natural arch' develops within the overlying roof strata. The

'natural arch' can be stabilised by reinforcing the rock in the loose zone below the arch. The Norwegian Institute for Rock Blasting Techniques has proposed a formula for determining the optimum bolt length required for the central section of such an opening, Equation 9 (3):

$$L = 1.40 + 0.184\ a \qquad \text{Eq. 9}$$

where L = Bolt Length (m)
a = Span of Opening (m)

If a natural arch is assumed to develop across the 7 West crosscut, due to the presence of an elevated horizontal regional stress field at Prince Mine (15), the optimum bolt length is:

$$L = 1.4 + 0.184\ (4.27) = 2.18\ m$$

In a reinforced arch, the excavation is bounded by a zone under compression with high tangential/circumferential stresses. The rock bolts are therefore used to increase both the shear resistance of the joints and the normal stress across the joints. In general, the optimum L/S ratio of bolt length (L) to bolt spacing (S) should be close to 2. However, the bolt spacing will depend upon the joint spacing (e). The relationship $s \leq 3\ e$ can therefore be used to determine the bolt spacing. In addition, the bolt tension (T) should lie between the limits (3):

$$0.5B < T < 0.8\ B \qquad \text{Eq. 10}$$

where, T = Applied Bolt Tension
B = Load Bearing Capacity of Bolt

4) U.S. Design Methods

Bieniawski presents a series of rock bolt design procedures developed for use in U.S. coal mines (4).

Rock load height (h_t) is determined using Equation 11:

$$h_t = \left(\frac{100 - RMR}{100}\right) \times B \qquad \text{Eq. 11}$$

where, RMR = Rock Mass Rating

B = Entry Width or Roof Span

a) Mechanical Anchor Design:

$$\text{Bolt Length } L_b = \frac{h_t}{2} \qquad \text{Eq. 12}$$

$$\text{Bolt Spacing } S_b = \sqrt{\frac{C_b}{SF \times \gamma \times h_t}} \qquad \text{Eq. 13}$$

C_b = Bolt Capacity

SF = Factor of Safety

γ = Unit Weight of Rock

h_t = Rock Load Height

Using the 7 West crosscut conditions:

$$\text{Bolt Length } L_b = \frac{3.50}{2} = 1.75 \text{ m} \qquad \text{from Eq. 12}$$

$$S_b = \sqrt{\frac{89.28}{1.5 \times 25.1 \times 3.5}} = \sqrt{0.68} = 0.82 \text{ m} \qquad \text{from Eq. 13}$$

where, C_b = 10 tnf or 89.28 kN

γ = 25.1 kN/m^3

h_t = 3.5 m

b) Resin Bolts:

$$\text{Resin Length } L_r = \sqrt{\frac{\gamma B^2 h_t}{2\ \sigma_h}} \qquad \text{Eq. 14}$$

where, σ_h = Horizontal Stress acting on the roof arch

If σ_h is not known then,

$$\text{Resin Length } L_r = \sqrt{\frac{B^2 h_t}{300}} \qquad \text{Eq. 15}$$

$$\text{Bolt Spacing } S_b = \sqrt{\frac{C_b}{SF \times \gamma \times h_t}} \qquad \text{Eq. 13}$$

Using the 7 West crosscut conditions:

$$\text{Resin Length } L_r = \sqrt{\frac{4.27^2 \times 3.5}{300}} = 0.46 \text{ m} \qquad \text{from Eq. 15}$$

$$\text{Bolt Spacing } S_b = \sqrt{\frac{89.28}{1.5 \times 25.1 \times 3.5}} = 0.82 \text{ m} \qquad \text{from Eq. 13}$$

DISCUSSION

The Q-System was originally developed from the analysis of over 200 tunnelling case histories collected in Scandinavia and it is unlikely that the data base contained any information collected in Coal Measures strata. The direct application of the Q-System to the Sydney Coalfield is uncertain and it is therefore necessary to assume that the behaviour of Coal Measures strata is directly comparable to the behaviour and physical properties of the weaker and altered rock types encountered in igneous and metamorphic formations.

Coal Measures strata normally contains well developed horizontal bedding features on which can be superimposed a joint pattern. However, the joint pattern is normally a secondary controlling feature when compared to those found in hardrock formations. It is therefore difficult to assign values to and determine the role of the Q-System parameters Jn, Jr, Ja and Jw for Coal Measures strata. In addition, advance longwall mining methods induce significant fracture patterns onto the surrounding rock mass and it is uncertain whether these can be directly compared to the joint networks encountered in hardrock operations.

Table 6 summarizes the results of the 7 West crosscut data obtained from the eight rock bolt design methods reviewed. The U.S. Corps of Engineers Option 1 is thought to be unrealistic with its recommendation of

		Bolt Length (m)	Bolt Spacing (m)
EMPIRICAL METHODS			
1) U.S. Corps of Engineers	Option 1	11.0	-
	Option 2	2.14	-
	Option 3	-	1.07
	Option 4	-	2.00
2) Geomechanics System	Best Option - Patterned grouted bolts on 0.75 m spacing with 100 mm thick shotcrete.		
	Worse Option - Stabilise with rope cover support and massive concrete 450 mm thick if stress changes not excessive.		
3) Q-System	Best Option - Tensioned bolts on 1 m spacing with mesh reinforced shotcrete 2.5 to 5 cm thick.		
	Worse Option - Mesh reinforced shotcrete 7.5 to 10 cms thick.		
4) French System		1.4 - 2.1	0.8
ANALYTICAL METHODS			
1) Bedded Strata - Competent Unit		4.0 - 4.5	1.0
2) Bedded Strata	Option 1	1.6	1.0
	Option 2	0.65	1.0
3) Arch Concept		2.18	-
4) U.S. Design Method			
	Case 1 - Mechanical Anchor Bolts	1.75	0.82
	Case 2 - Resin Anchor Bolts	0.46	0.82

Table 6 - Summarised results from the eight rock bolt design methods used on the 7 West crosscut conditions

11 m bolts because of operational considerations. Similarly, the support systems proposed by the Geomechanics and Q-Systems are also thought to be unrealistic because of their requirement for shotcrete. Both systems were derived from the analysis of data collected primarily from civil engineering projects where the requirements called for long-term permanent rather than short-term temporary support. It is interesting to note that based on the Geomechanics and Q-System analyses, the existing 5, 6 and 7 West roadways could not be supported using the rectangular steel sets (6W25 steel beams with 6W15 steel legs on 0.9 m spacings) currently employed.

The analytical methods indicate that bolt lengths between 0.65 m and 2.18 m could be used. However, the 4 to 4.5 m lengths required to anchor the mudstone to the overlying sandstone are thought to be unrealistic in terms of operational requirements.

The Q-System indicates that the optimum unsupported span for the 7 West crosscut conditions varies between 1.35 and 2.25 m using Equation 4 and 1.2 and 2.0 m using Equation 6. These values might be used to determine the optimum bolt spacing, although they are greater than the 0.82 and 1.0 m values given in Table 6.

Using the bedded strata with an overlying competent unit concept, the required capacity would be 19 tons for a 4 to 4.5 m bolt. However, using the bedded strata concept, bolts tensioned to 8 tons (80 kN) would be sufficient for bolt lengths of 0.65 or 1.6 m.

Three tentative bolting configurations are presented for use in the 7 West crosscut conditions, Figure 10. Figure 10(a), Case 1, shows an array consisting of five bolts, three on 1 m spacings across the roadway and one each inclined over the ribside. Figure 10(b), Case 2, shows an array consisting of cable slings. Figure 10(c), Case 3, shows a combination of roof bolts and cable slings. To prevent the spalling of loose rock from the roof, steel mesh should also be installed in conjunction with the final bolting configuration. In each case, the bolting system must be installed at the development face as soon after excavation as possible.

The use of rock bolts as a primary support mechanism would constitute a demonstration program. It is therefore suggested that normal rectangular steel sets also be installed in the trial crosscut as a secondary support system in case failures occurred and/or modifications were required to the original bolting pattern. A comprehensive geotechnical program should

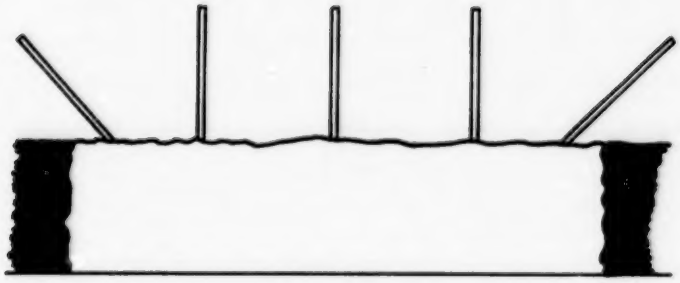

Figure 10 (a) Case 1 - Five rock bolts

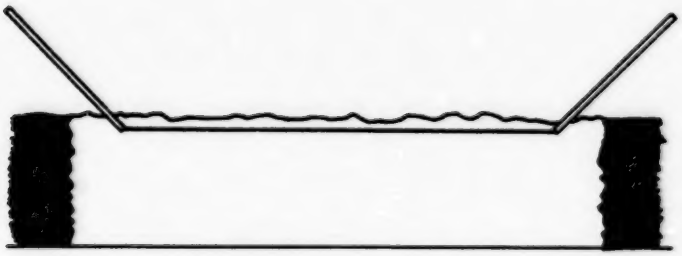

Figure 10 (b) Case 2 - Cable sling

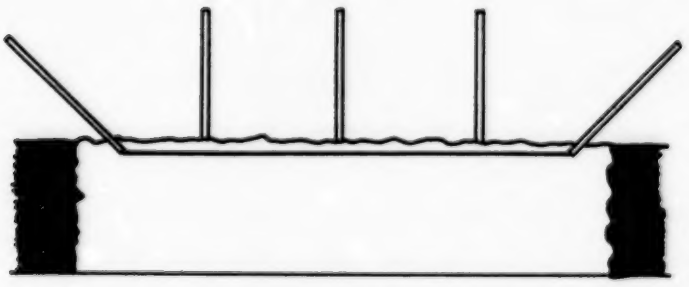

Figure 10 (c) Case 3 - Cable sling and three rock bolts

Figure 10 - Three bolting configurations proposed for the 7 West crosscut, Prince Mine

also be undertaken during the development and support of the crosscut. The program would involve the installation and monitoring of extensometer and convergence stations as well as conducting pull tests on selected bolts. The data obtained would be used to make 'real-time' alterations to the support system if required and optimise bolting configurations for future sites.

CONCLUSION

Rock bolting techniques are currently undergoing consideration as primary/secondary support systems for use in both temporary and permanent roadways within the Sydney Coalfield.

Data was input to eight different rock bolt design methods to compare the results and determine an optimum rock bolting system for use in a proposed 7 West crosscut at the Prince Mine. However, a wide variation in potential bolt lengths, spacings and capacities was obtained along with several unrealistic support methods involving the use of shotcrete and/or reinforced concrete.

Although no definitive bolting system was obtained, the results allowed three possible options to be formulated. However, specific details regarding these options and a final choice cannot be made until operational requirements/restrictions have been considered.

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APPENDIX A

Suggested Support Measures for the 38 Categories given in Figure 8 (after (3,8))

Support Measures for Rock Masses of "Exceptional", "Extremely Good", "Very Good" and "Good" Quality (Q range: 1000—10).

Support category	Conditional factors $\frac{RQD}{J_n}$	$\frac{J_r}{J_a}$	$\frac{\text{Span}}{ESR}$	Type of support	Note
1*	—	—	—	sb (utg)	—
2*	—	—	—	sb (utg)	—
3*	—	—	—	sb (utg)	—
4*	—	—	—	sb (utg)	—
5*	—	—	—	sb (utg)	—
6*	—	—	—	sb (utg)	—
7*	—	—	—	sb (utg)	—
8*	—	—	—	sb (utg)	—
9	⩾20	—	—	sb (utg)	—
	<20	—	—	B (utg) 2.5—3 m	—
10	⩾30	—	—	B (utg) 2—3 m	—
	<30	—	—	B (utg) 1.5—2 m + clm	—
11*	⩾30	—	—	B (tg) 2—3 m	—
	<30	—	—	B (tg) 1.5—2 m + clm	—
12*	⩾30	—	—	B (tg) 2—3 m	—
	<30	—	—	B (tg) 1.5—2 m + clm	—

Key to Support Tables:

sb = spot bolting
B = systematic bolting
(utg) = untensioned, grouted
(tg) = tensioned, (expanding shell type for competent rock masses, grouted post-tensioned in very poor quality rock masses; see Note XI)
S = shotcrete
(mr) = mesh reinforced
clm = chain link mesh
CCA = cast concrete arch
(sr) = steel reinforced

Bolt spacings are given in metres (m). Shotcrete, or cast concrete arch thickness is given in centimetres (cm).

Support category	Conditional factors $\frac{RQD}{J_n}$	$\frac{J_r}{J_a}$	Span / ESR	Type of support	Note
13	≥10	≥1.5	—	sb (utg)	I
	≥10	<1.5	—	B (utg) 1.5—2 m	I
	<10	≥1.5	—	B (utg) 1.5—2 m	I
	<10	<1.5	—	B (utg) 1.5—2 m + S 2—3 cm	I
14	≥10	—	≥15 m	B (tg) 1.5—2 m + clm	I, II
	<10	—	≥15 m	B (tg) 1.5—2 m + S (mr) 5—10 cm	I, II
	—	—	<15 m	B (utg) 1.5—2 m + clm	I, III
15	>10	—	—	B (tg) 1.5—2 m + clm	I, II, IV
	≤10	—	—	B (tg) 1.5—2 m + S(mr) 5—10 cm	I, II, IV
16* See note XII	>15	—	—	B (tg) 1.5—2 m + clm	I, V, VI
	≤15	—	—	B (tg) 1.5—2 m + S (mr) 10—15 cm	I, V, VI

* Authors' estimates of support. Insufficient case records available for reliable estimation of support requirements.

Note: The type of support to be used in categories 1 to 8 will depend on the blasting technique. Smooth wall blasting and thorough barring-down may remove the need for support. Rough-wall blasting may result in the need for single applications of shotcrete, especially where the excavation height is >25 m. Future case records should differentiate categories 1 to 8.

Support Measures for Rock Masses of "Fair" and "Poor" Quality (Q range: 10-1).

Support category	Conditional factors $\frac{RQD}{J_n}$	$\frac{J_r}{J_a}$	$\frac{\text{Span}}{ESR}$	Type of support	Note
17	>30	—	—	sb (utg)	I
	(≥10, ≤30)	—	—	B (utg) 1—1.5 m	I
	<10	—	≥6 m	B (utg) 1—1.5 m + S 2—3 cm	I
	<10	—	<6 m	S 2—3 cm	I
18	>5	—	≥10 m	B (tg) 1—1.5 m + clm	I, III
	>5	—	<10 m	B (utg) 1—1.5 m + clm	I
	≤5	—	≥10 m	B (tg) 1—1.5 m + S 2—3 cm	I, III
	≤5	—	<10 m	B (utg) 1—1.5 m + S 2—3 cm	I
19	—	—	≥20 m	B (tg) 1—2 m + S (mr) 10—15 cm	I, II, IV
	—	—	<20 m	B (tg) 1—1.5 m + S (mr) 5—10 cm	I, II

Key to Support Tables:

sb = spot bolting
B = systematic bolting
(utg) = untensioned, grouted
(tg) = tensioned, (expanding shell type for competent rock masses, grouted post-tensioned in very poor quality rock masses; see Note XI)
S = shotcrete
(mr) = mesh reinforced
clm = chain link mesh
CCA = cast concrete arch
(sr) = steel reinforced

Bolt spacings are given in metres (m). Shotcrete, or cast concrete arch thickness is given in centimetres (cm).

Support category	Conditional factors $\frac{RQD}{J_n}$	$\frac{J_r}{J_a}$	Span / ESR	Type of support	Note
20* See note XII	—	—	≥35 m	B (tg) 1—2 m + S (mr) 20—25 cm	I, V, VI
	—	—	<35 m	B (tg) 1—2 m + S (mr) 10—20 cm	I, II, IV
21	≥12.5	≤0.75	—	B (utg) 1 m + S 2—3 cm	I
	<12.5	≤0.75	—	S 2.5—5 cm	I
	—	>0.75	—	B (utg) 1 m	I
22	(>10, <30)	>1.0	—	B (utg) 1 m + clm	I
	≤10	>1.0	—	S 2.5—7.5 cm	I
	<30	≤1.0	—	B (utg) 1 m + S (mr) 2.5—5 cm	I
	≥30	—	—	B (utg) 1 m	I
23	—	—	≥15 m	B (tg) 1—1.5 m + S (mr) 10—15 cm	I, II, IV, VII
	—	—	<15 m	B (utg) 1—1.5 m + S (mr) 5—10 cm	I
24* See note XII	—	—	≥30 m	B (tg) 1—1.5 m + S (mr) 15—30 cm	I, V, VI
	—	—	<30 m	B (tg) 1—1.5 m + S (mr) 10—15 cm	I, II, IV

* Authors' estimates of support. Insufficient case records available for reliable estimation of support requirements.

Support Measures for Rock Masses of "Very poor" Quality (Q range: 1.0—0.1)

Support category	Conditional factors $\frac{RQD}{J_n}$	$\frac{J_r}{J_a}$	Span / ESR	Type of support	Note
25	>10	>0.5	—	B (utg) 1 m + mr or clm	I
	≤10	>0.5	—	B (utg) 1 m + S (mr) 5 cm	I
	—	≤0.5	—	B (tg) 1 m + S (mr) 5 cm	I
26	—	—	—	B (tg) 1 m + S (mr) 5—7.5 cm	VIII, X, XI
	—	—	—	B (utg) 1 m + S 2.5—5 cm	I, IX
27	—	—	≥12 m	B (tg) 1 m + S (mr) 7.5—10 cm	I, IX
	—	—	<12 m	B (utg) 1 m + S (mr) 5—7.5 cm	I, IX
	—	—	>12 m	CCA 20—40 cm + B (tg) 1 m	VIII, X, XI
	—	—	<12 m	S (mr) 10—20 cm + B (tg) 1 m	VIII, X, XI

Key to Support Tables:

sb = spot bolting
B = systematic bolting
(utg) = untensioned, grouted
(tg) = tensioned, (expanding shell type for competent rock masses, grouted post-tensioned in very poor quality rock masses; see Note XI)
S = shotcrete
(mr) = mesh reinforced
clm = chain link mesh
CCA = cast concrete arch
(sr) = steel reinforced

Bolt spacings are given in metres (m). Shotcrete, or cast concrete arch thickness is given in centimetres (cm).

Support category	Conditional factors $\frac{RQD}{J_n}$	$\frac{J_r}{J_a}$	Span / ESR	Type of support	Note
28*	—	—	≥30 m	B (tg) 1 m + S (mr) 30—40 cm	I, IV, V, IX
	—	—	(≥20 m, <30 m)	B (tg) 1 m + S (mr) 20—30 cm	I, II, IV, IX
See note XII	—	—	<20 m	B (tg) 1 m + S (mr) 15—20 cm	I, II, IX
	—	—	—	CCA (sr) 30—100 cm + B (tg) 1 m	IV, VIII, X, XI
29*	>5	>0.25	—	B (utg) 1 m + S 2—3 cm	—
	≤5	>0.25	—	B (utg) 1 m + S (mr) 5 cm	—
	—	≤0.25	—	B (tg) 1 m + S (mr) 5 cm	—
30	≥5	—	—	B (tg) 1 m + S 2.5—5 cm	IX
	<5	—	—	S (mr) 5—7.5 cm	IX
	—	—	—	B (tg) 1 m + S (mr) 5—7.5 cm	VIII, X, XI
31	>4	—	—	B (tg) 1 m + S (mr) 5—12.5 cm	IX
	≤4, ≥1.5	—	—	S (mr) 7.5—25 cm	IX
	<1.5	—	—	CCA 20—40 cm + B (tg) 1 m	IX
	—	—	—	CCA (sr) 30—50 cm + B (tg) 1 m	VII, X, XI
32	—	—	≥20 m	B (tg) 1 m + S (mr) 40—60 cm	II, IV, IX
See note XII	—	—	<20 m	B (tg) 1 m + S (mr) 20—40 cm	III, IV, IX
	—	—	—	CCA (sr) 40—120 cm + B (tg) 1 m	IV, VIII, X, XI

* Authors' estimates of support. Insufficient case records available for confident prediction of support requirements.

Support Measures for Rock Masses of "Extremely Poor" and "Exceptionally Poor" Quality (Q range: 0.1—0.001)

Support category	Conditional factors $\frac{RQD}{J_n}$	$\frac{J_r}{J_a}$	$\frac{\text{Span}}{ESR}$	Type of support	Note
33*	≥2	—	—	B (tg) 1 m + S (mr) 2.5—5 cm	IX
	<2	—	—	S (mr) 5—10 cm	IX
	—	—	—	S (mr) 7.5—15 cm	VIII, X
34	≥2	≥0.25	—	B (tg) 1 m + S (mr) 5—7.5 cm	IX
	<2	≥0.25	—	S (mr) 7.5—15 cm	IX
	—	<0.25	—	S (mr) 15—25 cm	IX
	—	—	—	CCA (sr) 20—60 cm + B (tg) 1 m	VIII, X XI
35	—	—	≥15 m	B (tg) 1 m + S (mr) 30—100 cm	II, IX
See note XII	—	—	≥15 m	CCA (sr) 60—200 cm + B (tg) 1 m	VIII, X, XI, II
	—	—	<15 m	B (tg) 1 m + S (mr) 20—75 cm	IX, III
	—	—	<15 m	CCA (sr) 40—150 cm + B (tg) 1 m	VIII, X, XI, III
36*	—	—	—	S (mr) 10—20 cm	IX
	—	—	—	S (mr) 10—20 cm + B (tg) 0.5—1.0 m	VIII, X, XI
37	—	—	—	S (mr) 20—60 cm	IX
	—	—	—	S (mr) 20—60 cm + B (tg) 0.5—1.0 m	VIII, X, XI
38	—	—	≥10 m	CCA (sr) 100—300 cm	IX
	—	—	≥10 m	CCA (sr) 100—300 cm + B (tg) 1 m	VIII, X, II, XI
See note XIII	—	—	<10 m	S (mr) 70—200 cm	IX
	—	—	<10 m	S (mr) 70—200 cm + B (tg) 1 m	VIII, X, III, XI

* Authors' estimates of support. Insufficient case records available for confident prediction of support requirements.

Supplementary notes by BARTON, LIEN *and* LUNDE

I. For cases of heavy bursting or "popping", tensioned bolts with enlarged bearing plates often used, with spacing of about 1 m (occasionally down to 0.8 m). Final support when "popping" activity ceases.

II. Several bolt lengths often used in same excavation, i.e. 3, 5 and 7 m.

III. Several bolt lengths often used in same excavation, i.e. 2, 3 and 4 m.

IV. Tensioned cable anchors often used to supplement bolt support pressures. Typical spacing 2—4 m.

V. Several bolt lengths often used in same excavation, i.e. 6, 8 and 10 m.

VI. Tensioned cable anchors often used to supplement bolt support pressures. Typical spacing 4—6 m.

VII. Several of the older generation power stations in this category employ systematic or spot bolting with areas of chain link mesh, and a free span concrete arch roof (25—40 cm) as permanent support.

VIII. Cases involving swelling, for instance montmorillonite clay (with access of water). Room for expansion behind the support is used in cases of heavy swelling. Drainage measures are used where possible.

IX. Cases not involving swelling clay or squeezing rock.

X. Cases involving squeezing rock. Heavy rigid support is generally used as permanent support.

XI. According to the authors' experience, in cases of swelling or squeezing, the temporary support required before concrete (or shotcrete) arches are formed may consist of bolting (tensioned shell-expansion type) if the value of RQD/J_n is sufficiently high (i.e. >1.5), possibly combined with shotcrete. If the rock mass is very heavily jointed or crushed (i.e. RQD/J_n <1.5, for example a "sugar cube" shear zone in quartzite), then the temporary support may consist of up to several applications of shotcrete. Systematic boltjng (tensioned) may be added after casting the concrete, but it may not be effective when RQD/J_n <1.5 or when a lot of clay is present, unless the bolts are grouted before tensioning. A sufficient length of anchored bolt might also be obtained using quick setting resin anchors in these extremely poor quality rock-masses. Serious occurrences of swelling and/or squeezing rock may require that the concrete arches are taken right up to the face, possibly using a shield as temporary shuttering. Temporary support of the working face may also be required in these cases.

XII. For reasons of safety the multiple drift method will often be needed during excavation and supporting of roof arch. Categories 16, 20, 24, 28, 32, 35 (SPAN/ESR >15 m only).

XIII. Multiple drift method usually needed during excavation and support of arch, walls and floor in cases of heavy squeezing. Category 38 (SPAN/ESR > 10 m only).

Supplementary notes by HOEK *and* BROWN *(1980)*

A. Chainlink mesh is sometimes used to catch small pieces of rock which can become loose with time. It should be attached to the rock at intervals of between 1 and 1.5 m and short grouted pins can be used between bolts. Galvanised chainlink mesh should be used where it is intended to be permanent, e.g. in an underground powerhouse.

B. Weldmesh, consisting of steel wires set on a square pattern and welded at each intersection, should be used for the reinforcement of shotcrete since it allows easy access of the shotcrete to the rock. Chainlink mesh should never be used for this purpose since the shotcrete cannot penetrate all the spaces between the wires and air pockets are formed with consequent rusting of the wire. When choosing weldmesh, it is important that the mesh can be handled by one or two men working from the top of a high-lift vehicle and hence the mesh should not be too heavy. Typically, 4.2 mm wires set at 100 mm intervals (designated 100 x 100 x 4.2 weldmesh) are used for reinforcing shotcrete.

C. In poorer quality rock, the use of untensioned grouted dowels as recommended by BARTON, LIEN and *Lunde* depends upon immediate installation of these reinforcing elements behind the face. This depends upon integrating the support drilling and installation into the drill-blast-muck cycle and many non-Scandinavian contractors are not prepared to consider this system. When it is impossible to ensure that untensioned grouted dowels are going to be installed immediately behind the face, consideration should be given to using tensioned rockbolts which can be grouted at a later stage. This ensures that support is available during the critical excavation stage.

D. Many contractors would consider that a 200 mm thick cast concrete arch is too difficult to construct because there is not enough room between the shutter and the surrounding rock to permit easy access for pouring concrete and placing vibrators. The US Army Corps of Engineers (1978) suggests 10 inches (254 mm) as a normal minimum while some contractors prefer 300 mm.

E. BARTON, LIEN and LUNDE suggest shotcrete thicknesses of up to 2 m. This would require many separate applications and many contractors would regard shotcrete thicknesses of this magnitude as both impractical and uneconomic, preferring to cast concrete arches instead. A strong argument in favour of shotcrete is that it can be placed very close to the face and hence can be used to provide early support in poor quality rock masses. Many contractors would argue that a 50 to 100 mm layer is generally sufficient for this purpose, particularly when used in conjunction with tensioned rockbolts as indicated by BARTON, LIEN and LUNDE, and that the placing of a cast concrete lining at a later stage would be a more effective way to tackle the problem. Obviously, the final choice will depend upon the unit rates for concreting and shotcreting offered by the contractor and, if shotcrete is cheaper, upon a practical demonstration by the contractor that he can actually place shotcrete to this thickness.
In North America, the use of concrete or shotcrete linings of up to 2 m thick would be considered unusual and a combination of heavy steel stets and concrete would normally be used to achieve the high support pressures required in very poor ground.

Supplementary note by STILLBORG

Untensioned, grouted rockbolts are recommended in several support categories. At the time when BARTON et al. proposed their guide for support measures the friction anchored rockbolts were not yet available. The note under Table 6 in connection with BIENIAWSKI's guide for excavation and support in rock tunnels is therefore equally applicable here.